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# A Study of the Concentration of a Copper-Zinc Ore.

Kenneth L. M. Dodd

Dodd, K. L. M.

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A Thesis
Submitted to the Department of Mineral Dressing
in Partial Fulfillment of the
Requirements for the Degree of
Bachelor of Science in Metallurgical Engineering

MONTANA SCHOOL OF MINES

Butte, Montana

May, 1940

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# A STUDY OF THE CONCENTRATION OF A COPPER-ZINC ORE

The ore under investigation in this thesis is a copper-zinc ore which came from the state of New Mexico. The exact location is unobtainable.

No data is available concerning the geological conditions in which the ore is present, nor the actual extent of the deposit.

The purpose of the investigation was to determine the most effective means of concentration of the valuable minerals present in the ore. There were two methods of attack. Firstly, it was determined whether the ore could be concentrated effectively by gravity-concentration methods; these tests subsequently failed to produce encouraging results. Secondly, following the failure of the gravity-concentration tests, the ore was tested to determine whether it was amenable to concentration by flotation, which proved to be very successful.

The procedure followed in the various tests was similar to that as outlined in the United States Bureau of Mines publication, Report of Investigations 3328.

## Crushing and Sampling

Sixty pounds of the ore, which averaged approximately $1\frac{1}{2}$ inches, were first passed through a laboratory gyratory crusher. The product from this preliminary crushing, which averaged about 0.5 inches in size, was then passed through a set of rolls. Approximately 90% of the secondary crushing

product passed through a 20 mesh screen; the plus 20 mesh material was ground on a bucking board to pass through the 20 mesh screen. The minus 20 mesh material was thoroughly mixed with a Jones splitter, following which a 350 gram sample was taken and ground on a bucking board to pass through a 100 mesh screen. This sample was then rolled in an oil cloth about one hundred times to insure a thorough mix. This product was used for the quantitative analysis tests.

The ore was analyzed for copper, zinc, iron, and insol. One half gram samples were taken and three charges were run for a check. The results of the analysis are given in Table I.

Table I

Analysis of Head Sample of Ore

| | |
|---|---|
| Copper | 1.05 % |
| Zinc | 18.46 % |
| Iron | 10.40 % |
| Insol | 34.50 % |

## Wet Screen Analysis

A wet screen analysis was made on a 500 gram sample of the ore. The products from wet screening are more suitable for float and sink analysis because of the removal of the slimes from the ore. The ore was first screened on a 65 mesh screen. The undersize from this screening was then screened on a 200 mesh screen. After drying, all of the plus 200 mesh sizes were aggregated and a Tyler screen test was made with the use of the Tyler Ro-Tap machine. The minus 200 mesh material from both the wet screening and the Ro-Tap sizing were also aggregated. There was a loss of 5 grams in the wet screening and 2 grams during the Ro-Tap screening which amounted to approximately one per cent of the total weight of the original sample. Float and sink analyses were made on the various mesh sizes from the set screening test.

The results of the wet screen analysis are given in Table II.

Table II

Screen Analysis of Ore

| Mesh | Wt. | Wt. % | Cum. Wt. % |
|---|---|---|---|
| 28 | 31.0 | 6.29 | 6.29 |
| 35 | 63.0 | 12.78 | 19.07 |
| 48 | 72.0 | 14.61 | 33.68 |
| 65 | 63.0 | 12.78 | 46.46 |
| 100 | 58.0 | 11.76 | 58.22 |
| 150 | 43.0 | 8.72 | 66.94 |
| 200 | 40.0 | 8.12 | 75.06 |
| -200 | 123.0 | 24.94 | 100.00 |
| Total | 493.0 | 100.00 | 100.00 |

## Float and Sink Analyses

Float and sink analyses are particularly valuable in ascertaining the distribution of the minerals in the various mesh sizes, in determining the extent of the interlocking minerals, and in preparing suitable portions for microscopic work.[1]

Acetylene tetrabromide (Sp. g. 2.95) was used for all of the tests.

Sized ore aggregated from the wet screening tests were used for the float and sink analyses. For particles coarser than 48 mesh, the separation was made in a 250 cc. beaker; for finer particles a 300 ml. separatory funnel was used. An average sample for the tests weighed 50 grams. For the coarser sizes, the sample was placed in a beaker nearly filled with the heavy liquid. The minerals with a density greater than the fractionating medium sink, while those with a lighter density float. The float fraction was skimmed off with a wire-screen scoop, drained, and thoroughly washed with

1 - Report of Investigation 3328 - U. S. Bureau of Mines

carbon tetrachloride to remove the last traces of the heavy liquid. The washed product was then dried and weighed. Next, the heavy liquid was drained from the sink fraction which remained in the beaker. This product was also washed, dried, and weighed as before.

The technique for samples finer than 48 mesh was quite similar. A sample of approximately 50 grams in weight was also used. Separation was effected in a 300 ml. separatory funnel. The sink product was drawn off first and collected on a filter paper. The float product was then run onto a filter paper and both products were drained, washed with carbon tetrachloride, and dried as previously noted.

In general, two minerals of similar shape porosity may be separated by gravity concentration if their densities in water differ by 0.5 or more.

The results of the complete float and sink tests are given in Table III.

Table III

Float and Sink Analysis Tests

| Mesh Size | Weights | | |
|---|---|---|---|
| | Float | Sink | Total |
| plus 28 | 13.10 | 17.97 | 31.07 |
| " 35 | 25.31 | 37.53 | 62.84 |
| " 48 | 25.04 | 45.61 | 70.65 |
| " 65 | 23.93 | 38.32 | 62.25 |
| " 100 | 22.00 | 36.32 | 58.32 |
| " 150 | 17.70 | 24.43 | 42.13 |
| " 200 | 15.38 | 23.40 | 38.78 |
| Totals | 142.46 | 233.58 | 376.04 |

## Microscopic Study

A microscopic examination of most samples is used to determine the minerals present, their relative abundance, and their grain size. For a complete study of the opaque minerals, the sample is mounted in a bakelite briquette and examined by reflected light.[1]

For microscopic study of the ore, a bakelite briquette of the minus 35 plus 48 mesh sink product from the float and sink analyses was prepared. Bakelite is an ideal mounting for mineral specimens because after moulding it is both hard and polished well.

In the preparation of the briquette the following procedure was employed: Equal volumes of the ore and bakelite were thoroughly mixed and placed in the briquette mold. Over this mixture a calculated quantity of bakelite was placed so that it sompletely filled the mold. The mold was placed in the machine with the plunger in place to which a pressure of 6,000 pounds per square inch was given. The mold was heated to a temperature of approximately 130 degrees C. by means of an electrical heating element. After the desired temperature was reached, additional pressure was added until the total was about 10,000 pounds per square inch. The heating was discontinued at this point and the mold was allowed to cool slowly to permit an optimum set for the bakelite. The briquette was removed from the mold and trimmed on an emery wheel. A preliminary polishing of the mineral surface of the briquette was given with 600 carborundum on a glass plate to remove any minute pits that

1- Report of Investigation 3328 - U. S. Bureau of Mines

might be present. The specimen was now ready for a further and a complete polishing on an automatic polishing machine, using copper and lead laps. Suitable grinding media were added to the laps to perfect a plane surface on the briquette.

After complete polishing, the briquetted sample was viewed under a metallurgical microscope using reflected light. The sulphides of iron (pyrite), copper (chalcopyrite), and zinc (sphalerite), with a quartz gangue, were identified.

The above minerals all appeared in the free state as well as being somewhat interlocked with each other. They were very clean looking with the exception of the sphalerite which appeared as one of the darker varieties of this mineral, commonly called black jack.

Each product from the float and sink analyses was examined under the microscope to determine the approximate size of liberation of the ore. Care was taken to notice any interlocked particles. The sizes which appeared most favorable for a more complete size of liberation were the following mesh sizes: plus 48, plus 65, and plus 100. Quantitative analysis tests were made on the float products from these mesh sizes. The results of these tests are given in TableIV.

TableIV

Chemical Analysis of Float Products

| Mesh | % Cu | % Zn | % Fe |
|---|---|---|---|
| plus 48 | 0.15 | 2.40 | 2.90 |
| " 65 | 0.12 | 2.21 | 2.52 |
| " 100 | 0.11 | 2.02 | 2.50 |

## Results of Float and Sink Analyses

The results from the float and sink analyses indicated that the size of liberation of the ore was minus 65 mesh, plus 100 mesh. It was immediately concluded from this data that gravity concentration methods had to be eliminated for concentration of the ore. Sized mineral particles smaller that plus 65 mesh cannot be effectively separated by a tabling operation.

Concentration of the ore was effected by flotation, from which excellent results were obtained.

# CONCENTRATION BY FLOTATION

## Introductory

In all, only four flotation tests were made. The first two tests were made with 500 gram samples using a 500 gram, Denver, laboratory flotation cell. The primary purpose of these first tests was to obtain preliminary information concerning the kind and amount of reagents to be added to effect a maximum recovery of the valuable minerals in subsequent tests. However, since the recovery of the metallics in the ore seemed to be good, these tests were not discarded. Only rougher concentrates and tails were recovered.

The final two tests were made with a 600 gram Fagergren laboratory flotation cell. This machine is manufactured by the American Cyanamid Company. The rougher concentrates were re-run without the addition of further reagents in order to recover a cleaner concentrate.

Quantitative analyses were run on all of the products

from the four flotation tests. From these analyses it was possible to calculate the recoveries of the various minerals present in the ore.

## Purpose

The essential purpose of the flotation tests was to effect a maximum recovery of the sphalerite. It was found impossible to recover the chalcopyrite since it's presence in the ore partly served to act as an activator for the sphalerite, resulting in a concentrate high in zinc and low in copper.

## Grinding Procedure

Charges weighing either 500 or 600 grams, depending on which flotation cell was going to be used for each test, were ground in an Abbe type pebble mill in preparation for the flotation tests to follow.

It is usual to screen out the undersize before the first grinding. However, with a sulphide ore such as was being tested in this thesis, it is preferable to subject the sample to a brief grinding period of one minute before screening in order to develop fresh mineral surfaces.

The charge was placed in the mill to which was also added the required charge of flint pebbles. Water was added to obtain a suitable pulp density. The mill was closed and rotated for one minute for the first grinding period. The mill was then removed from the rolls and the undersize was screened off by wet screening through a screen of a desired mesh. The oversize was returned to the mill for further grinding, after which the undersize was again screened off.

The oversize was reground in stages until all of the charge passed through the desired mesh.

The usual time of grinding was one minute for the first grind followed by successive grindings of from five to eight minute periods.

The undersize from the grindings was filtered with the use of a suction filter. The filter cake was transferred to a pan and covered with water where it was in readiness for the flotation test which was to follow.

## Reagents

The reagents used in the tests varied only as to the kinds of collectors and frothers. One conditioning reagent was used throughout; it also served as a depressant for the purite.

After a general survey of various zinc ore tests, the following reagents were decided upon:

Conditioning Reagent
- Lime (CaO)..... 4 to 5 lb./ton

Depressant
- Lime (CaO)..... 4 to 5 lb./ton

Activator
- $CuSO_4$.....1.5 to 2 lb./ton

Collectors
- Xanthate Z-4..... 0.1 lb./ton
- Xanthate Z-5..... 0.1 to 0.2 lb./ton
- Sodium Aerofloat..... 0.2 lb./ton

Frothers
- Pine oil..... 0.03 lb./ton
- Cresylic acid..... 0.03 lb./ton

NaCN could not be used as a depressant for pyrite because its use for such a purpose would have been destroyed by the addition of $CuSO_4$.

## Sequence of Reagent Addition

Conditioning agents other than activators or depressants were added first. Following a brief period of agitation the activator was added. The depressant should be added before the activation agent, but the lime or conditioner served this purpose as well as being a conditioning agent. The collectors were added next, followed by the frothers.

## Flotation Practice

The procedure followed in the various tests can be outlined in the following steps:

1. The filter cake was placed in the flotation cell and enough water (distilled) was added to give a suitable pulp density. The flotation reagents were added according to the general sequence as outlined above. Distilled water was used in all of the tests. A pulp dilution of 4 of water to 1 of solids was maintained in the Denver cell; a pulp dilution of 2.75 of water to 1 of solids was used in the Fagergren cell.

2. "pH" determinations were made on the rougher pulps with the La Motte roulette pH comparator. These values ranged from 10.0 to 10.8. In general, sufficient lime should be added to give a pH value of between 8.5 and 10.0 in order that the conditions should be preper to insure a rapid and efficient recovery of the zinc.[1] However, this range was exceeded during three of the tests without any noticeable handicap to a good recovery of the zinc.

3. Rougher concentrates, high in zinc and low in copper,

1 - "Flotation Reagents - 1940" American Cyanamid Co.

were recovered. The iron percentages were mostly in the tails. No further treatment was given to the first two tests. The final two tests were re-run without the addition of further reagents to recover a cleaner concentrates of zinc.

4. The products from the flotation tests were dried, weighed, and screened. Tyler screen tests were made with the use of the Tyler Ro-Tap machine.

5. Quantitative analyses were made on the minus 100 mesh material of each product.

6. Microscopic examinations were conducted on the minus 200 plus 270 mesh material of the first two rougher concentrates only. Bakelite briquets were made and viewed under a metallurgical microscope to identify the minerals and relative amounts of each present. The results from this work served to effect a higher recovery of the zinc and a more complete depression of the pyrite in the two following tests.

## Flotation Tests

### Tests No. 1 and 2

500 gram samples were used for both tests; both were conducted in the 500 gram, Denver laboratory cell. These tests were primarily to obtain practice in conducting a flotation concentration. However, it was noticed that the recoveries in both cases appeared to be good, so quantitative analyses were made on the products. The only products collected in both tests were rougher concentrates and tails. The ore sample for both tests was ground to approximately 90% minus 65 mesh.

The results obtained were as follows:

Table V

Reagent Consumption

| Reagents Used | Test No. 1 | Test No. 2 |
|---|---|---|
| CaO | 5.0 lb./ton | 4.0 lb./ton |
| $CuSO_4$ | 2.0 " | 1.5 " |
| Z-5 | 0.1 " | |
| Sod. Aero. | | 0.2 " |
| pine oil | 0.03 " | |
| cres. acid | | 0.03 " |
| pH | 10.4 | 10.0 |

Table VI

Recoveries

Test No. 1

| Product | Wt. Gms. | Wt. % | Assay percent Fe | Cu | Zn | Percent of total Fe | Cu | Zn |
|---|---|---|---|---|---|---|---|---|
| R. Conc. | 227 | 46.3 | 7.25 | 1.55 | 41.3 | 27.3 | 56.4 | 85.9 |
| Tails | 263 | 53.7 | 16.65 | 1.00 | 4.08 | 72.7 | 43.6 | 14.1 |
| Composite | 490 | 100.0 | 12.31 | 1.255 | 22.34 | 100.0 | 100.0 | 100.0 |

Table VII

Test No. 2

| Product | Wt. Gm. | Wt. % | Assay percent Fe | Cu | Zn | Percent of total Fe | Cu | Zn |
|---|---|---|---|---|---|---|---|---|
| R. Conc. | 221 | 44.6 | 14.90 | 1.25 | 42.10 | 58.3 | 46.7 | 89.0 |
| Tails | 274 | 55.40 | 8.58 | 1.10 | 4.12 | 41.7 | 53.3 | 11.0 |
| Composite | 495 | 100.00 | 11.40 | 1.194 | 21.08 | 100.0 | 100.0 | 100.0 |

Tests No. 3 and 4

600 gram samples were used for both tests and these experiments were performed with the 600 gram Fagergren cell. The rougher concentrates were re-run without the addition of further reagents to recover a cleaner concentrate.

The ore sample for both tests was ground in an Abbe pebble mill to approximately 90% minus 100 mesh.

The results obtained were as follows:

Table ~~IX~~ VIII

Reagent Consumption

| Reagents used | Test No. 3 | Test No. 4 |
|---|---|---|
| CaO | 5.0 lb./ton | 5.5 lb./ton |
| $CuSO_4$ | 2.0 " | 2.0 " |
| Z-4 | 0.1 " | |
| Z-5 | | 0.2 " |
| pine oil | 0.03 " | 0.03 " |
| pH | 10.2 | 10.8 |

Table IX

Recoveries

Test No. 3

| Product | Wt. Gm. | Wt. % | Assay percent Fe | Cu | Zn | Percent of total Fe | Cu | Zn |
|---|---|---|---|---|---|---|---|---|
| C. Conc. | 207 | 35.1 | 7.54 | 1.55 | 47.90 | 21.22 | 49.4 | 80.75 |
| C. Tails | 66 | 11.2 | 16.28 | 2.60 | 18.60 | 14.63 | 26.4 | 10.00 |
| R. Comc. | 273 | 46.3 | 9.64 | 1.80 | 40.80 | 35.85 | 75.8 | 90.75 |
| Tails | 317 | 53.7 | 14.90 | 0.50 | 3.58 | 64.15 | 24.2 | 9.25 |
| Composite | 490 | 100.0 | 12.46 | 1.104 | 20.80 | 100.00 | 100.0 | 100.00 |

Table XI

Test No. 4

| Product | Wt. Gm. | Wt. % | Assay percent Fe | Cu | Zn | Percent of total Fe | Cu | Zn |
|---|---|---|---|---|---|---|---|---|
| C. Conc. | 177 | 29.6 | 3.62 | 1.55 | 49.40 | 10.70 | 36.6 | 74.7 |
| C. Tails | 103 | 17.24 | 16.95 | 2.90 | 22.70 | 23.90 | 39.9 | 20.5 |
| R. Conc. | 280 | 46.84 | 8.53 | 2.05 | 39.60 | 34.60 | 76.5 | 95.2 |
| Tails | 318 | 53.16 | 14.90 | 0.55 | 1.74 | 65.40 | 23.5 | 4.8 |
| Composite | 598 | 100.00 | 12.23 | 1.251 | 19.145 | 100.00 | 100.0 | 100.0 |

## Discussion of the Results

As noticed in the tabulated results above, the maximum recovery of the zinc was obtained in Test No. 4 with a recovery of 95.2% zinc.

It was found impossible to effect a high depression of the chalcopyrite. Test No. 2 showed the highest depression of the copper with 53.3% of the total being removed in the tails.

Further experimentation with varying amounts of reagents similar to those used in Tests No. 2 and 4 would probably have resulted in a higher recovery of the zinc in the concentrates and a greater amount of the copper to the tails. Unfortunately, time did not permit further tests with these facts in mind.

Xanthate Z-5, as used in Test No. 4, showed the best collecting properties for the recovery of the zinc of all the collecting reagents employed in the tests.

Test No. 1 showed the best depression of the pyrite with 72.7% of the total iron in the ore going to the tails. Test No. 4 approached this figure closer than did the remaining two tests.

Tests No. 1 and 4 had the highest pH values for the pulp. Apparently the increased alkalinity served very well to effect a good depression of the pyrite.

## Conclusions

1. A zinc concentrate containing a maximum of 49.4% zinc was recovered from the ore. The overall recovery of the zinc was in excess of 95%. It is quite probable that a concentrate containing even a higher percentage of zinc can be recovered.

2. As only a zinc concentrate was desired, a recovery of the copper was not attempted.

3. The tests showed that gravity concentration methods cannot be used for the recovery of the zinc. Flotation methods were used, which proved to be very successful.

4. Since there is no data available concerning the extent of the ore body it is impossible to state whether it would warrant the erection of a mill or not.

PLATE I

Flowsheet of Laboratory Concentration Tests

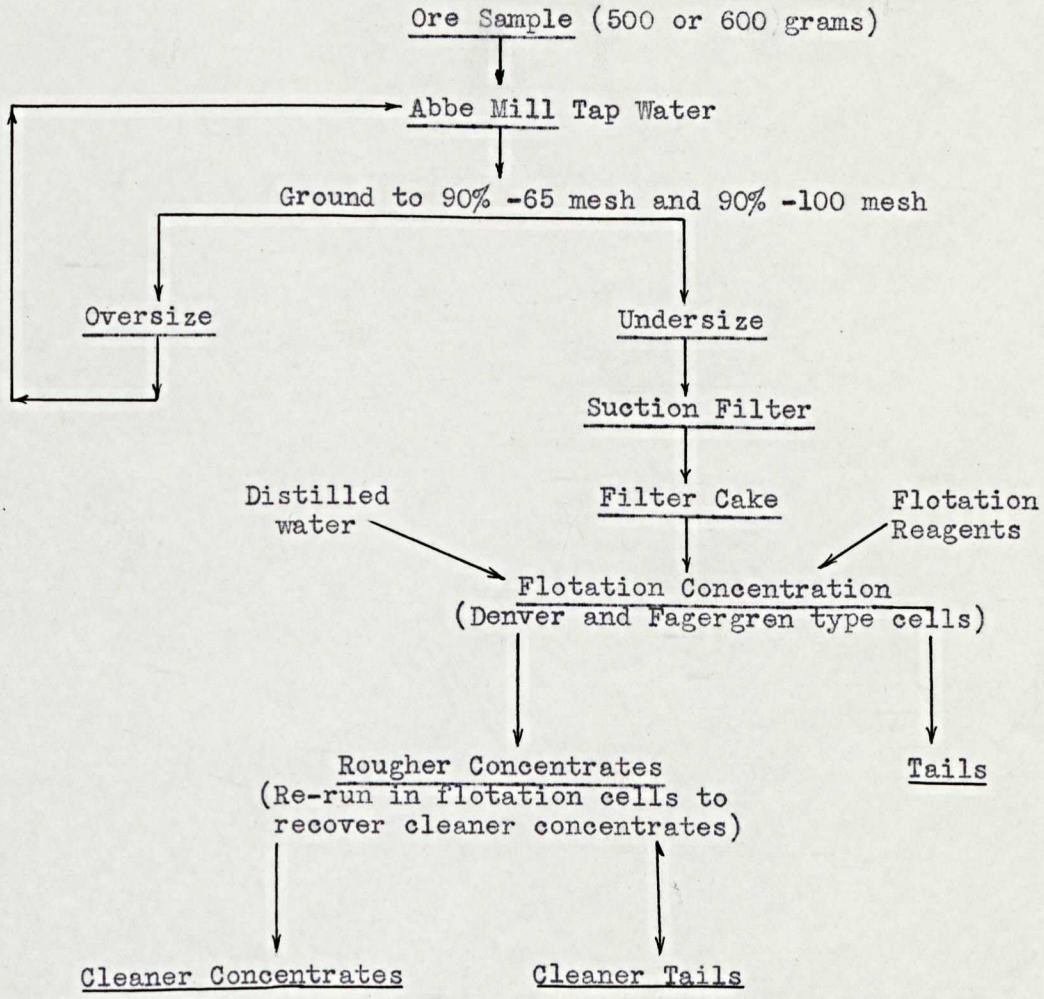
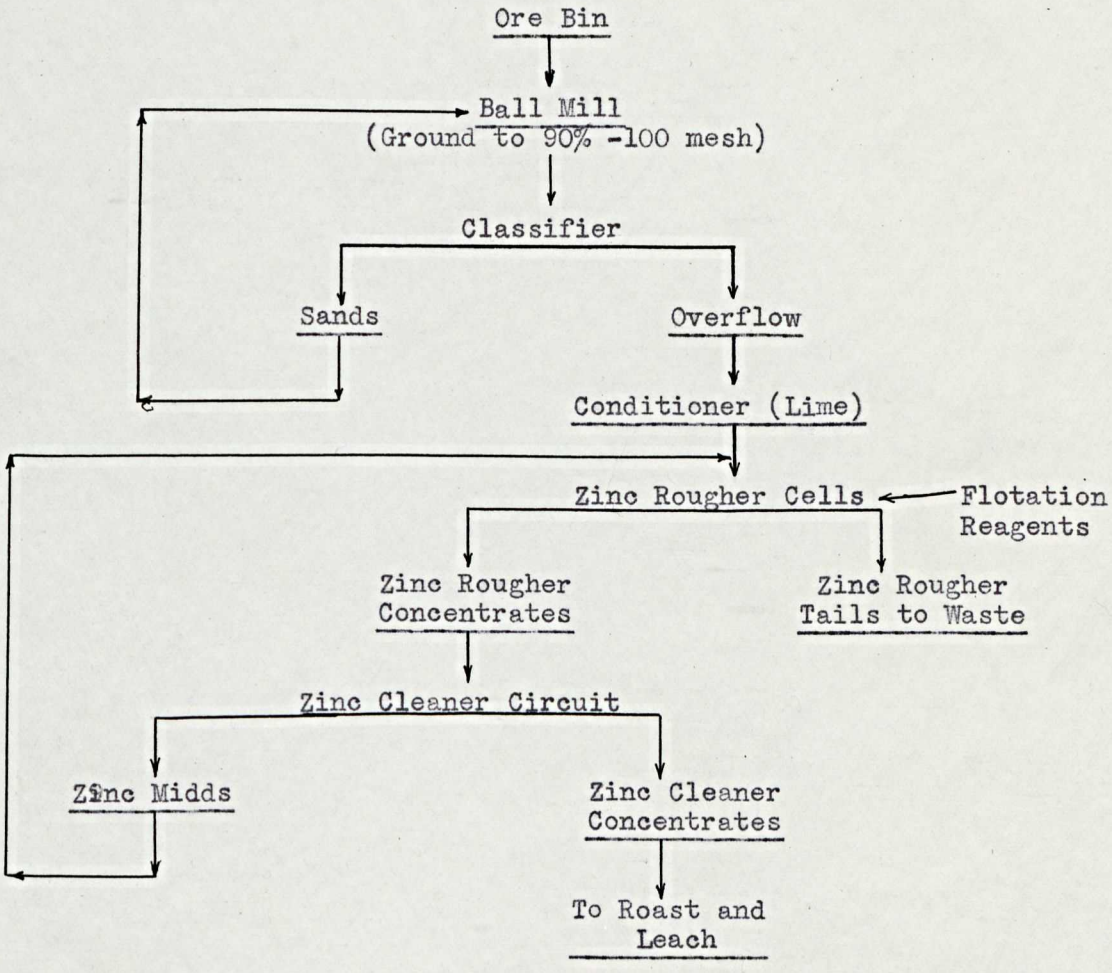


PLATE 2

Suggested Flowsheet for the Ore



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## Bibliography

1. "Report of Investigations--3328" United States Bureau of Mines.

2. "Flotation Reagents--1940" American Cyanamid Co.